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NEW SYSTEM OF MINING IN DZHEZKAZGAN

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[Tables and figures referred to herein are appended.]

The presence of cupriferous beds in Dzhezkazgan Rayon is indicated by mineralized arkosic sandstones cemented by quartz. Actually these sandstones occur on the surface and are oreless, or of shale and slate. The hardness of these sandstones according to the scale of Professor Protod'yakonov is 10 - 12. The shales are soft and therefore, for careful excavation under shale, a protective "back" layer of ore 1 - 1.5 meters thick is left to form the back or roof of the chambers. The dip of the beds is sloping and their thickness varies from 1.5 to 25 meters, averaging 8 meters.

Mining is done by the open-stope, room-and-pillar method. The chambers are 50 meters long and 20 meters wide. The interval between the pillar axes along the length of the chambers is 15 meters. Pillar diameter is 5 - 6 meters; thus, up to 14 percent of the general amount of ore is left in the pillars. Field training for students is carried out in new mines.

Although the mining-engineering conditions and methods of Dzhezkazgan are simple and well known to the world, Soviet engineers persistently worked on the development and utilization of new methods of excavation.

In 1947, experiments were carried out on two new systems of mining: (1) overhead excavation with partial storing of broken ore; and (2) underhand cutting with properly formed steps and zigzag shaping of the advanced stope.

The task of the author of the present article is to set forth the first results of the introduction of these methods of mining.

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I. OVERHEAD STOPING WITH PARTIAL STORING OF BROKEN ORE

This method was proposed by Candidate of Technical Sciences V. N. Semevskiy and received the 2d prize in the 1946 competition for the best mining methods.

To conduct the experiments, chamber No 2 in the southern side of Mine No 31 was chosen. The ore-bearing bed varies here from 10 to 13 meters in thickness. Red sandstone tops this body of ore. The chamber is situated in the zone of a fold and therefore the ore, especially in the beginning of the chamber, is broken horizontally. During overhead mining with partial storage of broken ore, excavation is begun by excavating from the earth of the chamber an undercutting level about 2 meters deep and then by excavating a second level of ore called the "floor-roof." The general height of the undercut and of the floor-roof must guarantee such disposition of the broken ore during cutting of the stratum remaining overhead that the distance between the roof of the chamber and the stored broken ore be sufficient for comfortable work. The height of the floor-roof is determined by an elementary calculation.

In the roof of chamber No 2, as during excavation of other chambers, a protective layer of uncaved ore about 1.5 meters thick was left and therefore the accountable thickness of the bed was accepted as 8.5 and 11.5 meters. For these conditions, the general height of the undercut and floor-roof was fixed at 5 and 6 meters and the floor-roof was given a stepped form, that permitted working the bed, which was of unusual thickness without inconvenience.

For the first 32 meters, the undercutting level and floor-roof was taken out simultaneously by underhand stoping (Figure 1 a). In the last 18 meters, an undercutting level 2.5 meters in height was first taken out and then a floor-roof of the same height. In the last instance, during excavation of the floor-roof blast holes were bored into the roof of the undercutting level by telescoping drilling machine. (Figure 1 b).

In September of 1947, excavation of the floor-roof was completed and then overhead work was undertaken to excavate the remaining mass of unbroken ore (Figure 1 c).

The initial mass of broken ore came first from the roof of the chamber. In consequence of countersinking the walls and of crumbling the ore bulkheads between the worked-out sections, an accumulation of broken ore resulted which permitted further cutting to be carried on.

In the overhand stope, two steps 2.5 - 3 meters in height were formed. Initial boring was done with telescoping drilling machines, but afterwards the boring of vertical holes was avoided because the ceiling became uneven. Irregularities in the ceiling and erection of shoring resulted from breaking the ore. Furthermore, telescoping drilling was complicated by the fact that the wedge-shaped, hard-alloy blades were often broken from the drills during such drilling.

For horizontal boring, two rows of holes were drilled on each step. The interval between rows is 0.6 - 0.8 meters and between holes in a row is 1.2 - 1.3 meters and the depth of the holes is 2.5 - 5 meters. The holes were bored from extensible platforms by manually operated drills weighing 21 and 27 kilograms. The drills had a chisel-like head equipped with wedge-shaped blades of hard alloy.

The distance between the stored broken ore and the uncaved ceiling of the chamber was 2.5 - 3 meters instead of the calculated 2 meters. This is explained in that the height of the "floor-roof" was raised a little here and there during its excavation.

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The ore was broken out in large pieces, because the blast holes were generally to the prevailing direction of fracturing.

Workers' movements upon the accumulated broken ore mass and drilling from it did not prove difficult. Occasionally drilling operations were carried out from a plank with one end on the broken ore and the other upon a ladder leading against the mining step (Figure 2).

This simple contrivance facilitated boring from the side of the accumulated ore. Ore was loaded into the cars by hand.

Trimming of the ceiling was done regularly be means of a long pole made from aluminum pipe. Shaping the supporting pillars was in no way difficult.

Technical-economic figures for ore-cutting in chamber No 2 are set forth in Table 1.

The output of the worker attendants in the same chamber for the period from the 21 October - 30 November amounted to 4.1 cubic meters. This period required 197 work-shifts and 807 cubic meters of ore were loaded. At the same time, 519 boulders were drilled, for the crushing of which 54.5 kilograms of explosives were used; that is, 67 grams per cubic centimeter. In the secondary crushing using drillers 13 work-shifts were expended.

A comparison of figures for overhead stoping and for the underhand stoping usually employed in Dzhezkazgan is set forth in Table 2.

The dat in Table 2 demonstrate the superiority of overhead stoping to underhand stoping accepted in Dzhezkazgan. The output of a driller was increased 2.5 times and that of a worker attendant was increased 1.5 times; and the expenditure of explosives was reduced by 3.3.

The increase in productivity of the driller and the reduction of the expenditure of explosives was achieved by better conditions for drilling and cutting. The increase of output of a worker attendant is explained by the constant reserve mass of broken ore in the stope. A comparison of labor for preliminary cutting and for the s ndary crushing is given in Table 3.

Outlays for the secondary crushing constitute about 20 percent of the total outlay for crushing ore. With regard to the large amount of work in the secondary crushing, it appears that this is a consequence of breaking the ore and not an inherent defect of the overhead cutting method.

Average work-shift productivity of a driller during overhead excavation and secondary crushing may be taken as $18.3 \cdot 0.83 = 15.2$ cubic meters or 39.5 tons.

In chamber No 2 the output of a driller was somewhat lower during telescopic drilling than during drilling of horizontal blast holes and the yield of large pieces of ore was significantly less.

Reduction of excessively large pieces of ore during overhead excavation may be accomplished by use of telescopic drills and by simultaneous explosion of charges by use of detonating fuse. In addition, it is necessary to establish the optimum distance between blast holes.

During working of the chamber by overhand stoping, the undercutting level should be excavated with an unstepped face 2 meters in height. Then the remaining volume of the chamber should be worked out overhead in order to maximize the overall efficiency of excavating.

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The working-out of the undercutting level and of the "floor-roof" by underhand stoping, as done in chamber No 2, reduces the general effectiveness of excavation and may be justified only under special conditions. Such a method was employed in chamber No 2 in an effort to improve the quality of ore, because the seam of ore at the undercutting level was relatively poor.

At a meeting of workers of the Dzhezkazgan Combine it was decided to continue experiments and widen application of experience gained. Overhand stoping with partial storage of broken ore was begun in chambers 20 and 21 of mine No 31 in December 1947.

The economic effect of introducing overhead excavation with partial storage of broken ore may be fixed approximately by the following methods of calculation.

The average thickness of ore beds being worked by overhead stoping is about 10 meters; hence 80 percent of the available ore will be worked by overhead stoping and 20 percent through excavation of the undercutting level.

The work-shift productivity of a driller during undercutting of the chamber, according to Table 1, amounts to 8.1 cubic meters or 21 tons; and during overhead cutting, including secondary crushing, it amounts to 15.2 cubic meters or 39.5 tons.

Average work-shift productivity of a driller is

$$(0.2 \cdot 21) + (0.8 \cdot 39.5) = 35.8 \text{ tons}$$

Since average work-shift productivity of a driller during underhand cutting amounts to 7.75 cubic meters or 20 tons, there is a saving of one ton of ore for equivalent labor of the driller amounting to $(1:20) - (1:35.8) = 0.022$ man-shift.

The pay rate of a driller in category 8 in Dzhezkazgan is 40.32 rubles. The saving on drillers' pay per one ton of ore, allowing 18 percent of the charges to wages, amounts to $0.022 \cdot 40.32 \cdot 1.18 = 1.05$ rubles.

Work-shift productivity of worker attendants during underhand excavation is 6.9 tons and during overhead it is 10.6 tons. The saving of attendant labor per one ton of ore is $(1:6.9) - (1:10.6) = 0.05$ man-shift.

The pay rate of a worker attendant in category 6 in Dzhezkazgan is 31.66 rubles. The saving in attendants pay per ton of ore is $0.05 \cdot 31.66 \cdot 1.88 = 1.86$ rubles.

In chamber No 2 ore was loaded by hand; therefore, there is no data for determining savings through the use of mechanical scraper loading. The large ore fragments were a known obstacle to mechanical scraper loading in chamber No 2. There is no doubt, however, that ore in normal-sized pieces can be obtained by boring with telescopic drills.

Savings would probably be twice as great through mechanical scraper loading than hand loading and would, therefore, amount to 0.93 rubles per ton of ore.

The over-all saving of man power per ton of ore amounts to $1.05 + 0.93 = 1.98$ rubles.

The outlay of explosives used in overhead cutting and for secondary crushing is 133 grams per ton; the average outlay in Mine No 31 is 300 grams per ton. The price of 1 kilogram of ammonium nitrate is about 1.9 rubles.

The saving per ton of ore is $(0.3 - 0.133) \cdot 1.9 = 0.31$ rubles.

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Over-all saving in man power and materials per ton of ore amounts to 1.98+0.31 = 2.29 rubles.

Besides the significant saving possible, overhead excavation with partial accumulation of broken ore permits an increase in production of an enterprise for the same amount of work and equipment.

II. UNDERHAND EXCAVATION WITH CORRECTLY FORMED STEPS AND ZIGZAG SHAPING OF THE ADVANCING STOPE

Experiments with this method of mining (Figure 3) were carried on in chamber No 3 in the southern section of Mine No 31. Thickness of the bed is 6 meters; the height of the advancing stope is 2 meters, of the first step 3 meters, and of the lower step 1 meter; width of the chamber is 15 meters.

The advancing face was zigzag shaped, usually having three protruding portions with the angles being about 120 degrees. Much ore remained in the recesses of the zigzags when the angles were closer to 90 degrees, which increased the amount of work in cleaning off the face. The zigzags were not always formed accurately enough due to the tendency of the ore to break into large lumps of improper shape. In each zigzag one or two rows of blastholes were bored, depending upon the depth. Three blastholes were placed in a single row, along the vertical.

The holes were 2.5 - 3 meters in depth and 42 millimeters in diameter. Boring was done with manual drills from an extensible support platform. The drills were equipped with wedge plates of hard alloy. One row of vertical holes were bored in the upper step. The interval between holes was 1.2 - 1.3 meters. The bottom step was mainly bored with horizontal holes.

The upper step was usually quite visible and correctly traced. The bottom step, on the contrary, was usually clogged with broken ore and was difficult to judge. For a more accurate shaping of the face it is necessary to have 4 or 5 steps.

Technical-economic figures of cutting of ore in chamber No 3 are set forth in Table 4.

Ore was loaded by hand. Production of worker attendants for the work period from 1 September through 1 December amounted to 2.75 cubic meters per shift; for this period 538 man-shifts were used and 1,485 cubic meters of ore were loaded.

A comparison of figures for underhand cutting with correctly formed steps in chamber No 3 with the average figures for Mine No 31 is set forth in Table No 5.

With properly formed steps and zigzag-shaped stoping, productivity of drillers increased 10 percent and attendants, 3 percent; and the expenditure of explosives was reduced 14 percent in comparison with the average results in Mine No 31.

This confirms the worth of having properly formed steps for underhand stoping; however, overhead excavation is shown to be considerably more efficient.

[Appended tables and figures follow.]

[redacted]

Table 1. Indices Measuring the Breaking-Up of Ore in Chamber No 2.

Character of the Work	Crushed ore in cubic meters	Work in man-shifts	Total length of holes bored in meters	Production per shift of a driller: Length in meters	Production per shift of a driller: Volume in cubic meters
Excavation of the under-cut layer and floor-roof in height 6 meters by underhand stoping	1428	164	1880	11.5	8.7
Excavation of the under-cut layer in height 2.5 meters	1246	154	1700	11	8.1
Excavation of a floor-roof 2.5 meters in height by overhand stoping	176	8	120	15	22.0
Formation of the first pile of broken ore	531	43	586	13.6	12.4
Overhand-stoping excavation. Drilling by telescoping perforators	747	41	470	11.5	18.3
Overhand-stoping excavation. Drilling by horizontal drills	2147	98	1062	10.8	22.0

Table 1 (Contd)

Character of the Work	Expenditure per cubic meter: Explosives in grams	Expenditure per cubic meter: Drillings in decameters	Period of work
Excavation of the under-cut layer and floor-roof in height 6 meters by underhand stoping	810	13.2	1/II 16/VII
Excavation of the under-cut layer in height 2.5 meters	1000	13.6	16/VII 26/VIII
Excavation of a floor-roof 2.5 meters in height by overhand stoping	310	6.8	26/VIII 27/VIII
Formation of the first pile of broken ore	550	11.0	1/IX 20/IX

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Table 1 (Contd)

Character of the Work	Expenditure per cubic meter: Explosives in grams	Drillings in decameters	Period of work
Overhand-stoping excavation. Drilling by telescoping perforators	280	6.3	20/IX 10/X
Overhand-stoping excavation. Drilling by horizontal drills	233	4.9	10/X 13/XI

Table 2. Comparative Data of the Two Methods of Excavation

Index	Underhand-stoping excavation	Overhand-stoping excavation
Production per shift in cubic meters		
Driller	7.75	22.0
Worker attendant	2.66	4.1
Expenditure of explosives in grams per cubic meter	780	233

Table 3. Data on the Expenditure of Explosives and Drills During Initial Breaking of Ore and During Rebreaking

Operations	Expenditure of explosives: Grams per cubic meter	Percent	Expenditure of drills: Decameters per cubic meter	Percent
Initial overhand-stoping breaking	280	81	6.3	83
Rebreaking	67	19	1.3	17
Total	347	100	7.6	100

Table 4. Indices Measuring the Breaking of Ore in Chamber No 3

Character of the Work	Work in man-shifts	Length of holes drilled (total) in meters	Volume of ore broken in cubic meters	Production per shift of drillers: Linear meter	Cubic meter
Drilling of horizontal holes in the front stope face	92	1125	560	12.1	6.1
Drilling on the ledges of vertical holes and horizontal holes	83	1112	925	13.2	11
Average indices for the room	-	-	-	12.7	8.5

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Table 4 (Contd)

Character of the Work	Expenditure per cubic meter: Explosives in grams	Expenditure per cubic meter: Drilling in decameters	Period of work
Drilling of horizontal holes in the front stope face	720	20.0	1/IX 1/XII
Drilling on the ledges of vertical holes and horizontal holes	650	13.2	1/IX 1/XII
Average indices for the room	670	15.0	

Table 5. Comparative Data on the Breaking of Ore in Chamber No 3 and Mine No 31

Indices	Mine No 31	Room No 3
Production per shift in cubic meters		
Driller	7.75	8.5
Worker attendant	2.66	2.75
Expenditure of explosives in grams per cubic meter	780	670

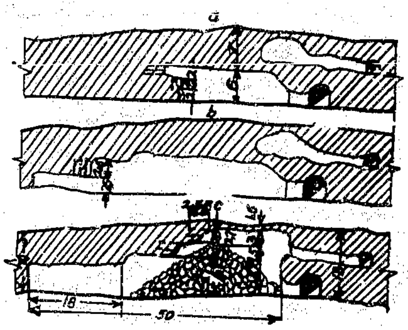

Figure 1. Successive Stages in Working Chamber No 2
a. Excavation of undercut and "floor-roof," working downwards (underhand stoping)
b. Excavation of "Floor-roof" overhead (overhand stoping)
c. Overhead excavation with partial storage of broken ore

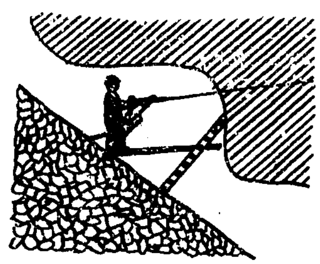

Figure 2. Boring From the Side of Accumulated Mass of Broken Ore

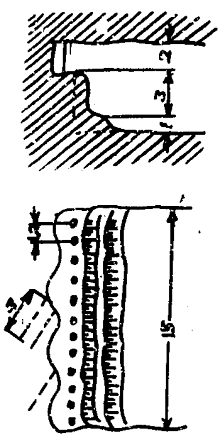

Figure 3. Scheme of Work in Chamber No 3

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